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REPORT ON BLAST DESIGN FOR NARROW VEIN STOPES LUPIN MINE

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43 pp

MICROMEDIA

ABSTRACT

The reported project was intended to introduce and refine a drill pattern for the West Zone of the Lupin Mine which would provide adequate breakage, yet reduce drilling and explosive use and thereby reduce wall damage and control dilution without introducing extensive artificial ground support. This zone is mined using a mechanized sublevel open stoping technique.

A review of current practices was undertaken by way of a literature search and direct communication with mines identified as using similar mining methods.

Current practices at Lupin were assessed and criteria for assessing blast performance from a production objectives and blast monitoring perspective were established. Blast patterns were designed based on current understanding and test blasts were undertaken to test the design.

The success of various patterns is reported, together with the conclusion that, for the established mining geometry, blasthole drilling inaccuracies will have to be addressed before significant beneficial results can be achieved from the proposed pattern changes.

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1.0 INTRODUCTION

A blast monitoring and design project was undertaken at the Lupin Mine in 1989 by Echo Bay Mines Ltd. with support from the Canada - NWT Mineral Development Agreement (MDA).

This project was undertaken to introduce and refine a drill pattern which will provide adequate breakage, yet reduce drilling and explosive use in the sublevel longhole open stopes of the narrow West Zone of the Lupin deposit. Expected changes would reduce wall damage and control dilution without introducing extensive artificial ground support.

Several factors that significantly affect blast performance are:

- hole alignment
- burden and spacing
- initiation method
- delay scatter
- explosive type and charge density
- blast hole interaction

These factors were studied and experimented with in order to optimize drilling and blasting.

The knowledge gained from this project will have potential application at other mines implementing longhole stoping in narrow deposits. Narrow vein longhole open stoping can be a safe, low cost, high productivity mining method if wall dilution can be minimized. This can be achieved through increased understanding and improved technology. The trend toward greater mechanization and the shortage of skilled shrinkage miners will give longhole methods increased significance in narrow ore bodies.

1.1 Scope

The initial steps of the project were to review the existing literature and accumulate an appropriate data base. This exercise was undertaken by Yves Lizotte of CANMET. There was little of relevance available in the published literature, but through personal communications and a questionnaire to appropriate organizations the current practices in narrow vein longhole open stoping of a number of mines have been documented. The results of this are reported in "Narrow Vein Blasthole Stoping: Current Drilling and Blasting Technology" by Y. Lizotte MRL89-93 (IR) which is incorporated as Appendix A of this report.

Monitoring, review and evaluation of current blasting practices at the Lupin Mine were undertaken, and is reported in Section 3.0 of this report. The available data was compiled and analyzed, with recommendations for optimization by Golder Associates Ltd.

Field trials of the recommended patterns were undertaken with blast monitoring using the methods described in Section 4.3.

Conclusions and recommendations regarding the major factors identified for blasting requirements in the West Zone at Lupin, improvements, and future developments are set out (Section 5.0) and potential application at other mines is discussed (Section 6.0).

2.0 MINE HISTORY

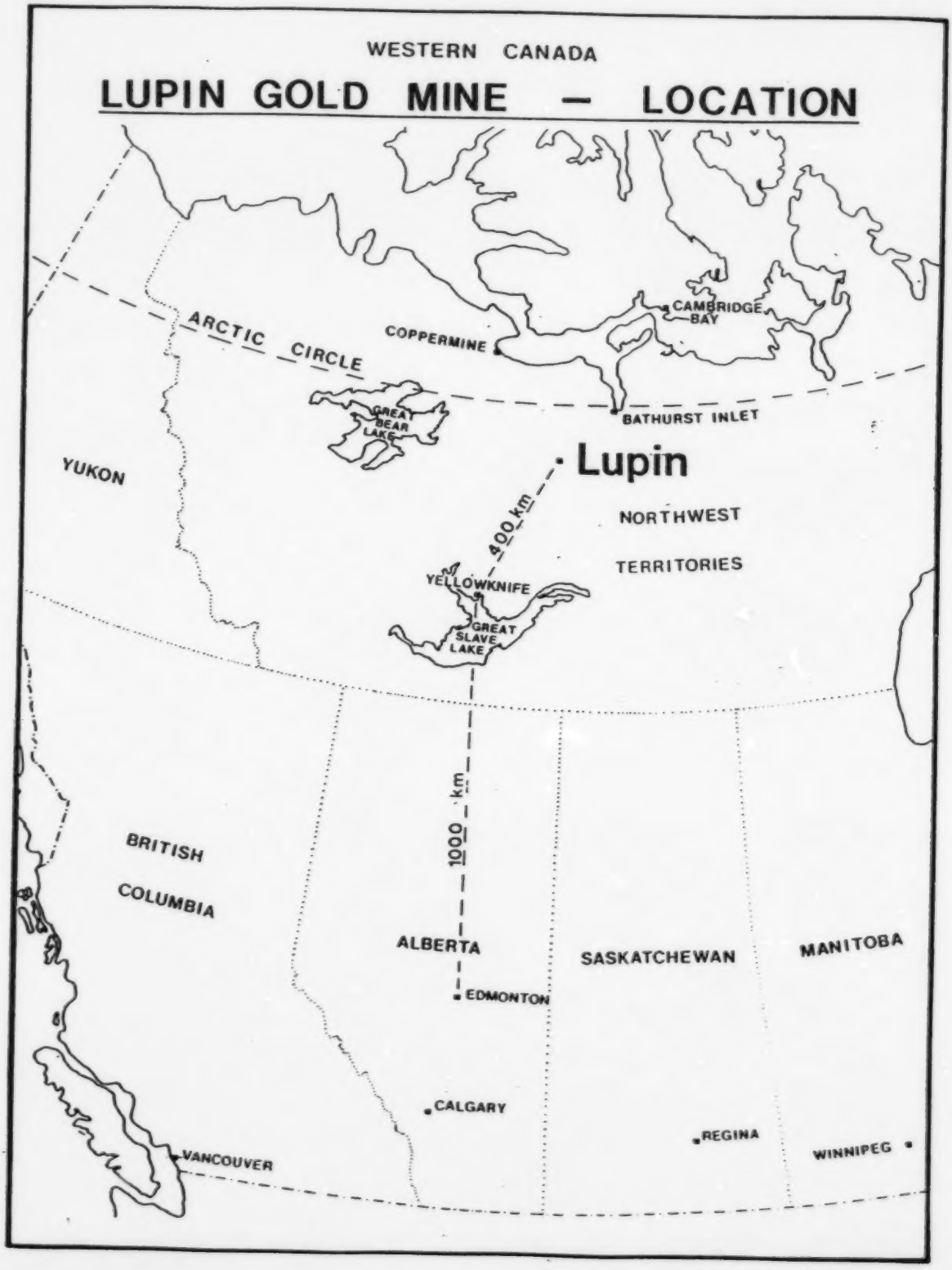
2.1 General

The Lupin Mine is a gold mining, milling and refining complex located 400 km northeast of Yellowknife on the southwest shore of Contwoyto Lake (see Figure 2-1). It currently produces 6000 kg of gold from 620,000 tonnes of ore each year.

The mine is in the continental sub-arctic climatic zone, with an annual mean temperature of -12°C. Permafrost extends from near surface to 500 m deep in the ore zone.

LUPIN GOLD MINE LOCATION

Figure **2-1**



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Production began in 1982 at 860 tonnes per day and has been increased over the years to the current rate of 1700 tonnes per day.

2.2 Geology

The Lupin gold deposit occurs in the sulphide rich portion of an amphibolitic iron formation which makes up part of the Contwoyto Formation sedimentary sequence. It is overlain by phyllites and underlain by quartzites. It has been folded and tilted into a tight anticline - syncline structure plunging to the Northeast at 75°.

In plan view the orebody appears as a Z-shaped structure which consists of three zones: the West Zone, Center Zone and East Zone (see Figure 2-2). The Center Zone has a strike length of greater than 200 m and is the widest ore zone, varying from 4.5 to 20 m width. The East Zone contains ore along up to 150 m of strike length, but no ore has been confirmed in that zone below the 350 m level. The West Zone is the narrowest ore zone with an average width of 1.5 m, but the structure extends more than 1000 m along strike (of which, typically, 70% would be ore grade material). The Center and West Zones have been confirmed at a depth of 890 m.

2.3 West Zone Mining

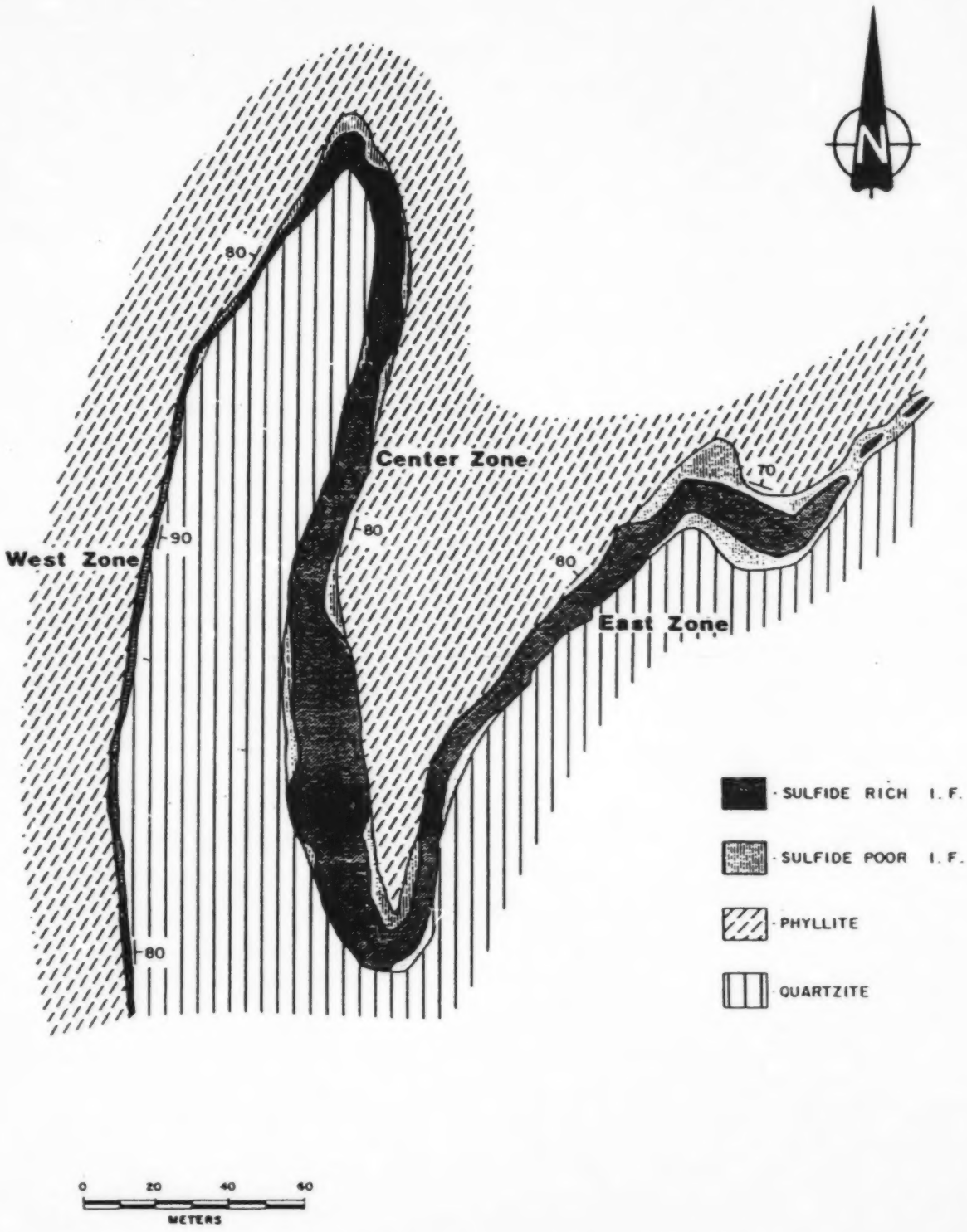
The current ore reserves for the West Zone are approximately 2.7 million tonnes. Several mining methods have been tried in the West Zone to recover this ore in a productive and cost efficient manner.

Initially, the equipment and methods used in the Center Zone were applied in the West Zone. This resulted in high dilution because the stope tended to fail to the width of the undercut subdrifts, regardless of the planned stope width.

In 1983 shrinkage stoping was introduced. This was an appropriate method for the orebody, but the cyclic nature of the method, limited productivity, industry wide lack of skilled workforce and high unit costs made it unattractive.

TYPICAL PLAN VIEW OF ORE ZONE

Figure **2-2**



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To address the productivity and dilution concerns, the raise platform mining (RPM) approach was introduced in 1986. This method consisted of driving parallel raises at 12 m centers through the ore zone for vertical distances of 80 m and then drilling off breast holes between the raises. The ore between the raises was blasted progressively from bottom to top and mucked from drawpoints and haulage drifts on the lower level. Better productivity was achieved with good dilution control, but the desired productivity levels were not achieved and problems of cyclic production, lack of skilled miners and some safety concerns remained.

Throughout these trials, portions of the West Zone continued to be mined using sublevel longhole open stoping. In 1987 a review of the method was conducted in order to address the specific problem areas that had been identified. This led to adoption of the sublevel longhole open stoping method currently in use. More than 100,000 tonnes of ore was mined from the West Zone by this method in 1989 and the production will double in 1990.

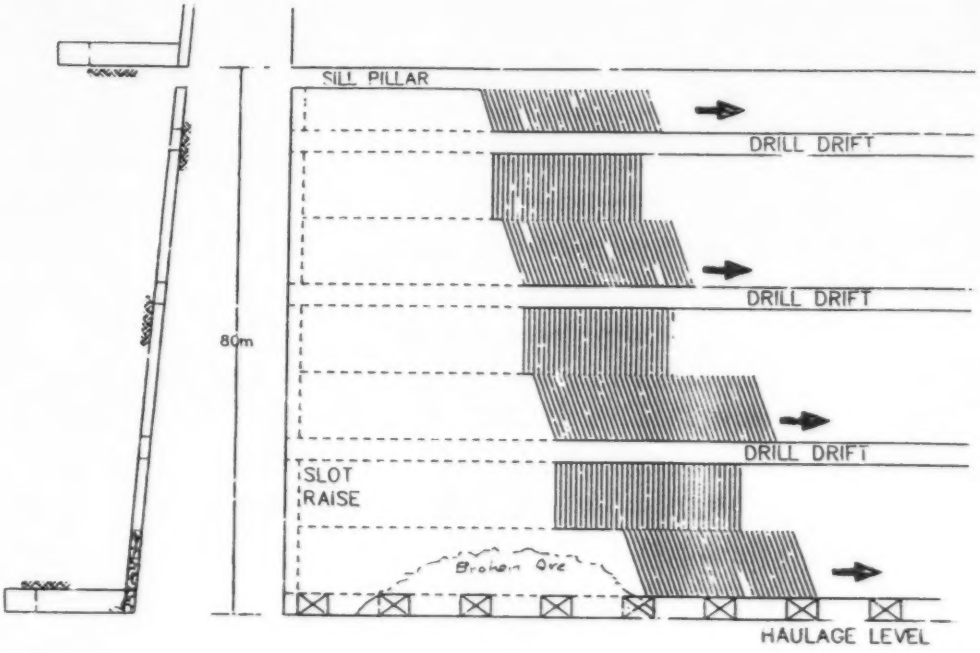
There is a main haulage level for every 80 m high block and the sublevels are on 20 m intervals. The ore drifts are driven 2.0 m wide by 3.0 m high to expose the full width of the ore zone. The production holes are 10 m long and are drilled down and up from each sub-level (except the haulage level which is upholes only) to reduce hole length and deviation (see Figure 2-3).

The upholes are dumped 70 degrees towards the stope to achieve a smooth back when blasting. A 50 mm diameter hole is used to drill a 3:2 pattern (Figure 3-1). The use of this pattern on a narrow ore body results in close hole spacing and a high powder factor. This causes over-blasting, but was implemented to minimize the potential for benches in blasting this narrow zone.

No artificial ground support is being used in these stopes and some sloughing occurs, particularly in the North end of the West zone where weak structures parallel the ore. Current drilling and blasting practice may contribute to this sloughing.

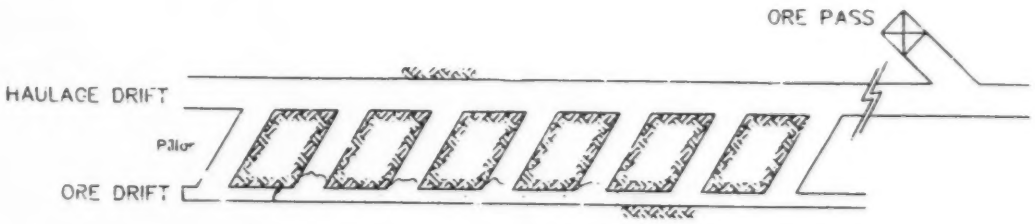
TYPICAL WEST ZONE LONGHOLE MINING

Figure **2-3**



CROSS SECTION

LONGITUDINAL SECTION



PLAN

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3.0 BLASTING PRACTICES

Standard procedures for production blasts at the Lupin Mine are outlined in this section.

3.1 Drilling

Production blastholes are drilled with two Tamrock H506 RTS Micro Solo longhole drills equipped with Tamrock HE 538 drifters. Holes are drilled 50 mm in diameter, and up to 12 meters long. This is done using 32 mm circular-section drill rods threaded both ends and connected using female/female couplings.

At the minimum stope width the 3:2 pattern drilled with a 0.75 m burden and a one metre overlap between down and up holes gives a drilling productivity of 1.32 tonnes per metre (t/m) drilled. At a two metre stope width the productivity is 1.76 t/m. Comparable productivities using a 0.65 burden and 2:1 pattern are 1.9 and 2.5 t/m respectively.

Holes are drilled in rows, with drill layout sections provided for each set of two rows. The layout specifies dip angle and direction, drill boom dump angle and direction, hole length and whether it is expected to be a breakthrough hole or not. The holes are drilled parallel wherever the stope geometry and drill limitations permit. To limit dilution, the holes are drilled to a toe target of 0.3 m inside the ore contours.

Hole lengths are designed with a 1.0 m overlap between stopes to reduce the possibility of benching. Downholes are drilled vertically and the upholes are drilled with the boom dumped 70 degrees towards the free face to limit benching at the toe of the hole. This design and the production cycle indicate that when blasted the downholes should be breakthrough holes while the upholes are not.

Collar accuracy is measured by inserting a close fitting plastic pipe in the hole and measuring dip angles, dump angles and collar location. A tolerance of $\pm$ 2° is allowed because of the precision of the tools being used. A satisfactory method of routinely measuring hole accuracy at the toe has not been found.

Where circumstances lead to drilling holes to breakthrough on the level below, strings weighted with various bolts and washers are lowered down the holes to identify which holes broke through. On occasion, the breakthrough locations are also surveyed. This surveying has indicated that holes are frequently up to 2 metres off plan along strike for an 18 m hole length.

3.2 Blast Preparation

When the blasters arrive to prepare the West Zone for a stope blast they first locate and clean the collars of all holes (based on the blaster's copy of the ring layout sections). They then measure the depth of the holes (down holes in particular). Ice is frequently encountered in downholes because these holes are not normally drilled to breakthrough, and because drill water left in the hole will freeze in place. If holes are frozen up some ANFO is poured into the holes to help thaw them. Alternatively, if the holes are plugged at a high level the drill is brought back into the stope to ream out the holes.

3.3 Loading and Initiation

The Blasthole Loading procedure is as follows:

1. The rings and holes to be blasted are located.

2. The holes are cleaned, measured, and downhole breakthrough is determined. Holes are marked for re-drilling if necessary.

3. Breakthrough downholes are plugged with one suspended stick of Powermex (with up to three additional sticks gravity loaded).

4. Downholes are pneumatically loaded with ANFO to a depth of approximately 4 m from the collar, primed (single holes are double primed to ensure detonation), then loading continues to the collar (no stemming).

5. Upholes are also pneumatically loaded to a depth of approximately 4 m from the collar, primed, and loaded to the collar (no stemming).

6. A Nonel MS Delay cap and Cilgel 70% booster cartridge is the primer in all holes.

7. Holes are connected using B-line.

All sequencing is at the discretion of the individual blaster. Single cap number delays are typically used between rows of holes (see Figure 3-1).

4.0 ASSESSMENT OF BLASTING PRACTICE

A summary of the production and development blasts monitored as part of this study is presented in Table 4-1. A detailed review of each blast is presented in Appendix B.

4.1 Production Criteria

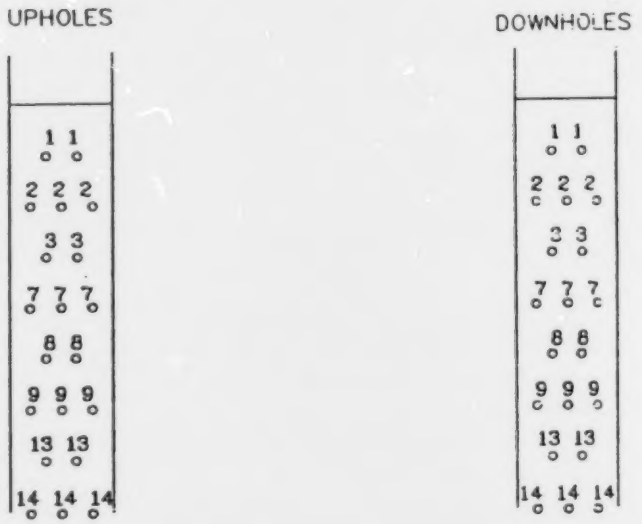
Certain objectives for mine operations provide criteria for assessing the blast performance with respect to production. These include:

- Adequate breakage - muck must be broken well enough to be readily loaded at the drawpoints with full bucketloads and without the necessity of secondary blasting in the drawpoints or plugging up the ore pass grizzly.

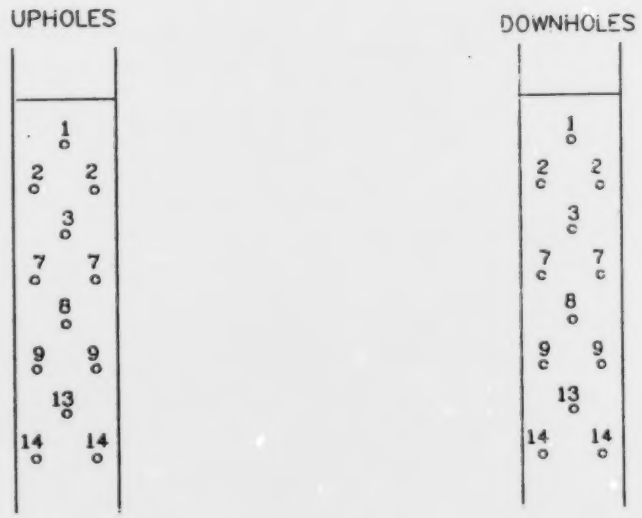
- Minimal dilution - overbreak into stope walls or sloughing from the walls causes dilution of the ore, thereby reducing ore grade and increasing the cost per unit of product.

- Minimal cost - only sufficient drilling and blasting should be done to achieve the required results. Over-drilling and over-blasting increases direct mining costs.

BLASTHOLE INITIATION SEQUENCE | Figure **3–1**



3:2 Pattern



2:1 Pattern

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- Minimize benching - if a blast fails for some reason and a remnant has to be re-accessed and re-blasted, or has to be abandoned and a new slot raise established, then production is disrupted and extra costs are incurred.

4.2 Historical Development of Blasthole Patterns

During the development and implementation of sub-level open stoping (SLOS) for extraction of the West Zone orebody a number of blasthole patterns were assessed. Initially, mining involved the use of Centre Zone equipment and techniques, then progressed to smaller drilling equipment and blasting practices more suitable to the narrow ore zone.

This use of this method (SLOS) is still evolving for narrow orebodies and specific problems concerning drilling accuracy must still be addressed.

4.2.1 2:1 Pattern, 63 mm Blasthole Diameter

Blasting this pattern in the last quarter of 1985 gave the following results (B. Schultz, 87/07/12):

> "While blasting this pattern through October, November and December the stope was carefully monitored. Initially the stope walls were in very good shape though minor localized areas slabbed due to wallrock cleavage. No evidence of undercutting failures was observed. Just north of the 67 level waste pillar, however, the stope began to break to a "slip" on the east wall about 1 metre outside the ore contour. Throughout the blasting the walls remained smooth and continuous. To prevent this type of overbreak occurring south of the waste pillar two measures were adopted. First, all perimeter drill holes were designed to collar and parallel the ore contour several inches inside the ore/waste contact. Second, swellex rockbolts were installed every 1.6 metres from the back of the drift extending into the footwall and hanging wall at angles of 45° and 60°. Initially the rockbolts straddled the ore/waste contact but later the bolts were installed just outside the contact".

"Pertinent observations in the rockbolted area revealed that initially the walls were smooth and breakage occurred to the designed width. Continued blasting again resulted in very smooth, uniform walls but 0.3 - 0.5 m of overbreak was suspected on each side of the ore contour. Recent blasting shows minimal overbreak and the walls are continuing to look good. The back is breaking properly, exposing shattermarks and the muck is very fine. Most rockbolts are still in place though several feet of each are exposed. Overbreak appears to have stabilized at about 0.3 m on either side".

"With positive results obtained from the 2:1 pattern the burden was expanded to 1.0 metre between rings on downholes and 0.94 metres between rings on upholes in December of 1985. The remaining 1800 metres of the 67 WZ stope was drilled off using this pattern, realizing over 7400 tonnes production at a tonnes per metre ratio of 4.15 with minimal drilled dilution. This pattern was maintained where the ore body was between 1.5 metres to 2.5 metres in width with extra holes being added where the ore width exceeded this margin. Perimeter drillholes were collared approximately 0.2 m inside the ore contours allowing for overbreak."

"Blasting began and was completed in the first two weeks of January, 1986 in this area. A small bench was encountered on six downhole rings - probably caused by insufficient amounts of explosive loaded in the holes."

4.2.2 2:1 Pattern, 50 mm Blasthole Diameter

Mining with 50 mm diameter blastholes began in April, 1988. Initially holes were drilled using a 2:1 pattern with a 0.75 m burden between rows (1.5 m between 2 hole rows).

Downhole blasts of 2-6 rings per blast benched at the toes of the holes on several occasions. Muck observed from these blasts was small where breakage occurred. Stope observations showed significant overbreaking of the footwall as well (C. Richards 88/05/11).

Due to the benching problem two additional holes were drilled in the one-hole row, converting to a 3:2 pattern.

During the assessment of this pattern, no hole deviation measurements were made and therefore the degree to which hole deviation influenced the problem of benching was never established.

4.2.3 3:2 Pattern, 50 mm Blasthole Diameter

This blasthole pattern (late 1988) represented a 40% increase in drilling over the previous 2:1 pattern. While this pattern reduced the occurrence of benches, the potential for blast induced damage was increased.

Two stopes have been mined to date using this pattern. However, significant wall sloughing was observed together with some oversize muck in drawpoints. This was particularly true at the north end of the WZ 170/87 stope block which broke to a graphite-coated surface.

4.3 Blast Monitoring Method

Blast vibration monitoring and design has traditionally involved measuring peak particle velocity and applying a charge weight per delay criteria. This method has been applied successfully in the far field where the blasts acts as a point source of energy. In the near field, however, the equipment normally associated with this method and the charge weight per delay criteria do not allow for a valid assessment of the blasting mechanism. The equipment typically used cannot sample at sufficient rates to separate individual holes detonating on the same delay period and, hence, explosive energy cannot be assigned to specific blastholes. This limits the degree to which a blast can be analyzed and optimized. The charge weight per delay criteria is not valid in the near field due to a most basic assumption that an explosive column detonates instantly at the exact nominal delay time. The criteria does not acknowledge the finite velocity of detonation (VOD) of an explosive product or the existence of scatter in the delay elements.

The limitations of traditional methods led to extensive research programs around the world. The most successful program was carried out at the Julius Kruttschnit Mineral Research Center (JKMRC), University of Queensland, Australia. Developments at the JKMRC included high speed digital recorders (BMX Blast Monitor), analysis software and detailed monitoring techniques. Golder Associates Ltd. (GAL) developed a working agreement with the JKMRC in 1987, and a transfer of this technology took place during the following year.

This new technology is centered around the following:

- installing low cost, high performance transducers within and behind the blast;
- recording the blast event at very high sampling rates using specially developed digital recorders (BMX);
- analyzing the event as a series of discrete energy signals (i.e., individual blastholes);
- implementing and monitoring incremental changes to blasting practices as a path towards optimization of these practices.

Details of the transducers and digital recorder are presented in Appendix C.

4.4 Blast Monitoring Criteria

In order to analyze the blast performance effectively, a set of criteria must be established to quantify performance. The assessment used in this study was based upon the following aspects:

- Efficiency and control with which all the explosive charges within a blast detonate.

Blasts can be quantified according to the efficiency of charge performance. At the Lupin Mine misfires and low efficiency detonations occurred at an average of 10 per cent. Until each charge effectively imparts its energy to the rock mass in a controlled manner, attempts to optimize blasting performance will have limited success.

- Size distribution of the muckpile

Examination of the muckpile revealed fragmentation to be predominantly fine in size, although a number of large pieces were generally present. Bi-model muck size distribution of this type is characteristic of overcharging, where the finer product comes from the "true" blasted rock and the large fragments are generated by overbreak and sloughing.

- Post-blast condition of the stope walls

Another means of assessing performance is to examine the rock that the blasting mechanism was meant to leave intact. This assessment involves visual examination of the stope walls and estimation of the ore dilution.

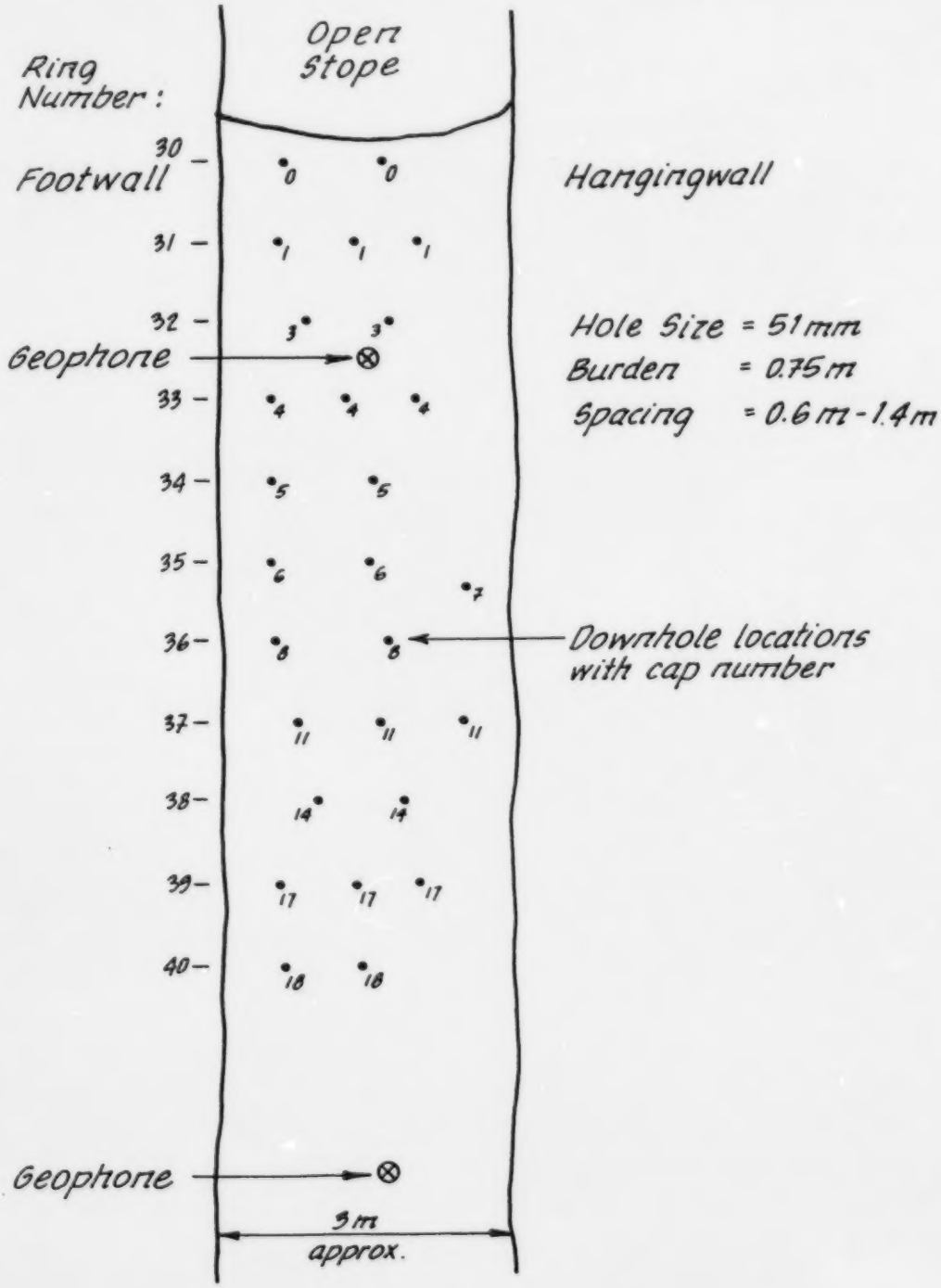
In the West Zone there was obvious wall sloughing and heavy dilution. Historical evidence (RPM Stopes) has shown that this does not always occur and therefore may be attributed to the blasting practice and hole deviation in association with footwall structure closely paralleling the ore zone.

4.5 Blast Monitoring Results

4.5.1 3:2 Pattern

A 3:2 pattern, shown in Figure 4-1 and discussed in 4.2.3, was monitored in the 130 WZ stope block. The stope was instrumented with five biaxial geophone arrays and four blasts were monitored.

DRILLHOLE PLAN FOR 130 WEST ZONE 3:2 PATTERN | Figure **4–1**



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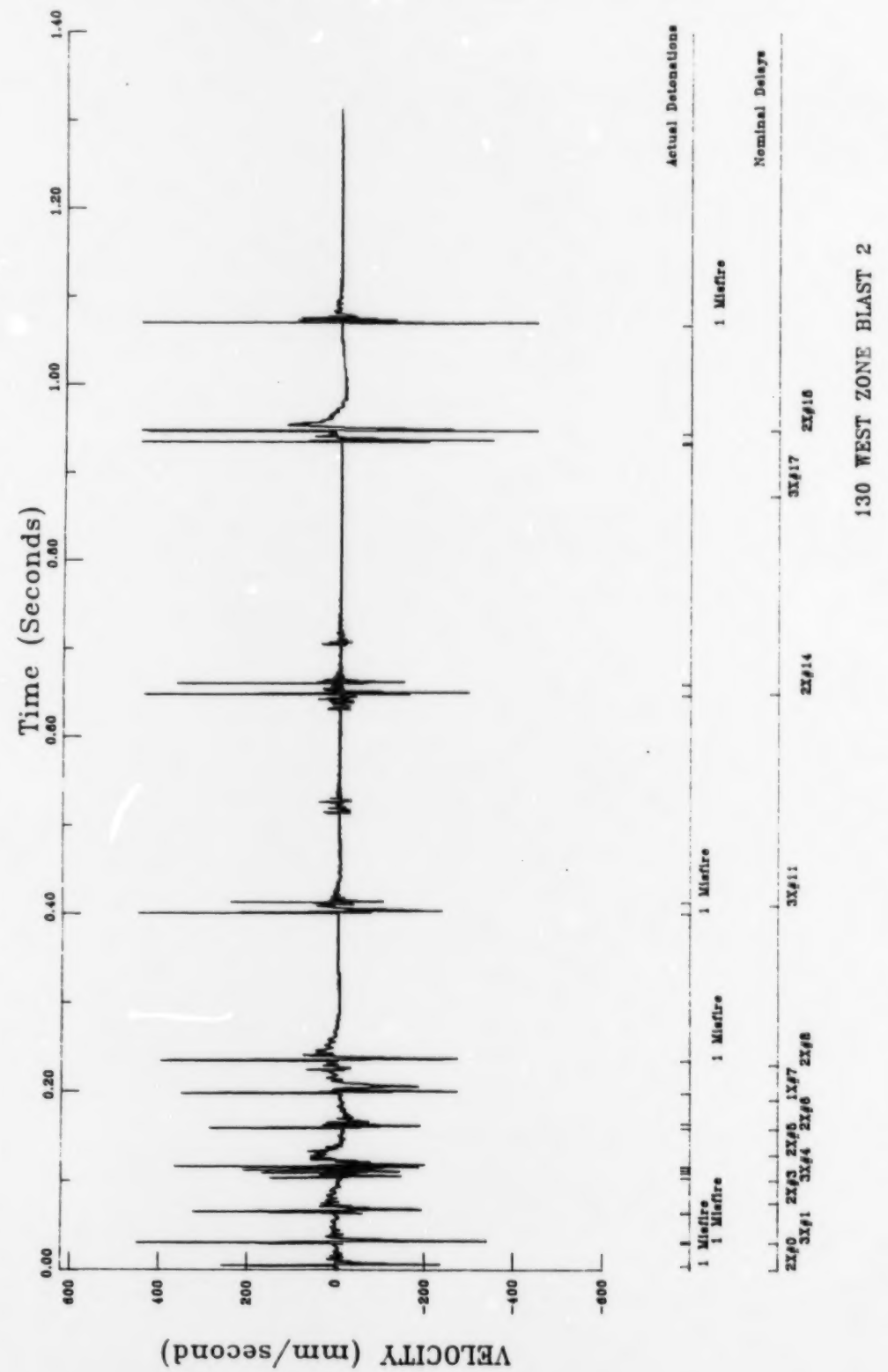
VIBRATION RECORD FOR FIGURE 4-1 BLAST

Figure **4-2**



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Monitoring identified the following:

- Significant charge interaction due to overbreak

  This resulted in substantial misfires and inefficient detonations.

  The mechanism was easily identified in the vibration record presented in Figure 4-2 for the #11 delay period. In this case, only one hole of a three hole ring detonated efficiently, one detonated inefficiently and one misfired. The first hole in the ring to fire moved its nominal burden, and a substantial portion of the remaining ring burden; this resulted in one dislodged charge and one totally displaced charge. This problem was evident in all 3:2 pattern blasts monitored.

- High energy levels at distances beyond the designed zone of influence of the blast.

  Energy levels in excess of 900 mm/s were measured at a distance of 3 m behind the blast. This value would also be representative of energy levels at a similar distance into the stope walls. The associated wall damage contributed to considerable dilution when the footwall failed to a prominent structural feature.

  High energy levels are often a result of poor sequencing techniques. When all three holes in one ring are on the same delay, the probability of a wall hole detonating prior to the center hole (and hence be overly confined) is 66 per cent. This increase in confinement resulted in increased tearing and shearing along the stope walls. As well, the greater the number of holes per delay, the greater the probability of significant energy enhancement.

- Significant backbreak of up to one third of the nominal burden.

This resulted from overcharging and potentially from inadequate uncharged hole lengths and lack of stemming.

This could lead to unsafe loading situations during the next cycle.

- Lower energy levels as a result of diluted ANFO.

  When a diluted ANFO (CIL's LOMEX II) was used, energy levels decreased by approximately 40 per cent with no evidence of degradation of the muck product.

4.5.2 2:1 Pattern (Modified 3:2 Pattern)

A 2:1 pattern was monitored in the 130 WZ stope block. The side holes of the 3:2 pattern were left uncharged effectively creating the 2:1 pattern. A single blast using this pattern was monitored.

Monitoring identified the following:

- An efficient blast mechanism

  All holes detonated and contributed to fragmentation of the rock mass. No misfires or inefficient detonations were recorded.

- No significant decrease in energy levels

  Energy levels recorded were consistent with those generated by the 3:2 pattern. Blast induced failure of the footwall still occurred despite a 40 per cent decrease in charge weight.

4.5.3 2:1 Pattern

A 2:1 pattern, shown in Figure 4-3, was monitored in the 270 WZ South stope block. The stope was instrumented, with one biaxial and one triaxial geophone array, and two blasts were recorded.

Monitoring identified the following:

- An efficient blasting mechanism

  Both blasts performed as designed with no recorded misfires.

  A typical vibration record is presented in Figure 4-4. This is a major step towards optimization. When every hole is efficiently contributing to breakage, the pattern (explosive distribution) can be modified further to increase blast efficiency.

- Potentially damaging energy being imparted into the stope walls

  Energy levels in excess of 900 mm/s were recorded at a distance of 4 m behind the blast.

  Similar energy levels could be expected at the same distance into the stope walls. However, the associated wall damage was much less severe than that attributed to the 3:2 pattern. An important note was that damage was restricted to the hangingwall and not associated with the faulting in the footwall.

  The energy levels were substantially higher than desired at this distance from the blastholes.

4.6 Blast Monitoring Discussion

The successful analysis of a blast design is dependent upon the quality of many fundamental blasting factors:

- blasthole drilling accuracy

  The analysis of any blast layout assumes that the blastholes are accurately drilled to their design location.

- explosive product and accessory performance

  The analysis assumes that charge malfunctions are a result of blast design rather than product failure.

- stope preparation and loading techniques

  The analysis assumes that charge malfunction are a result of blast design rather than poor quality workmanship.

This program did not assess the quality of the pre-blast factors, although it did become apparent that the above factors had a major influence upon the outcome of the blasts (and hence the monitoring program).

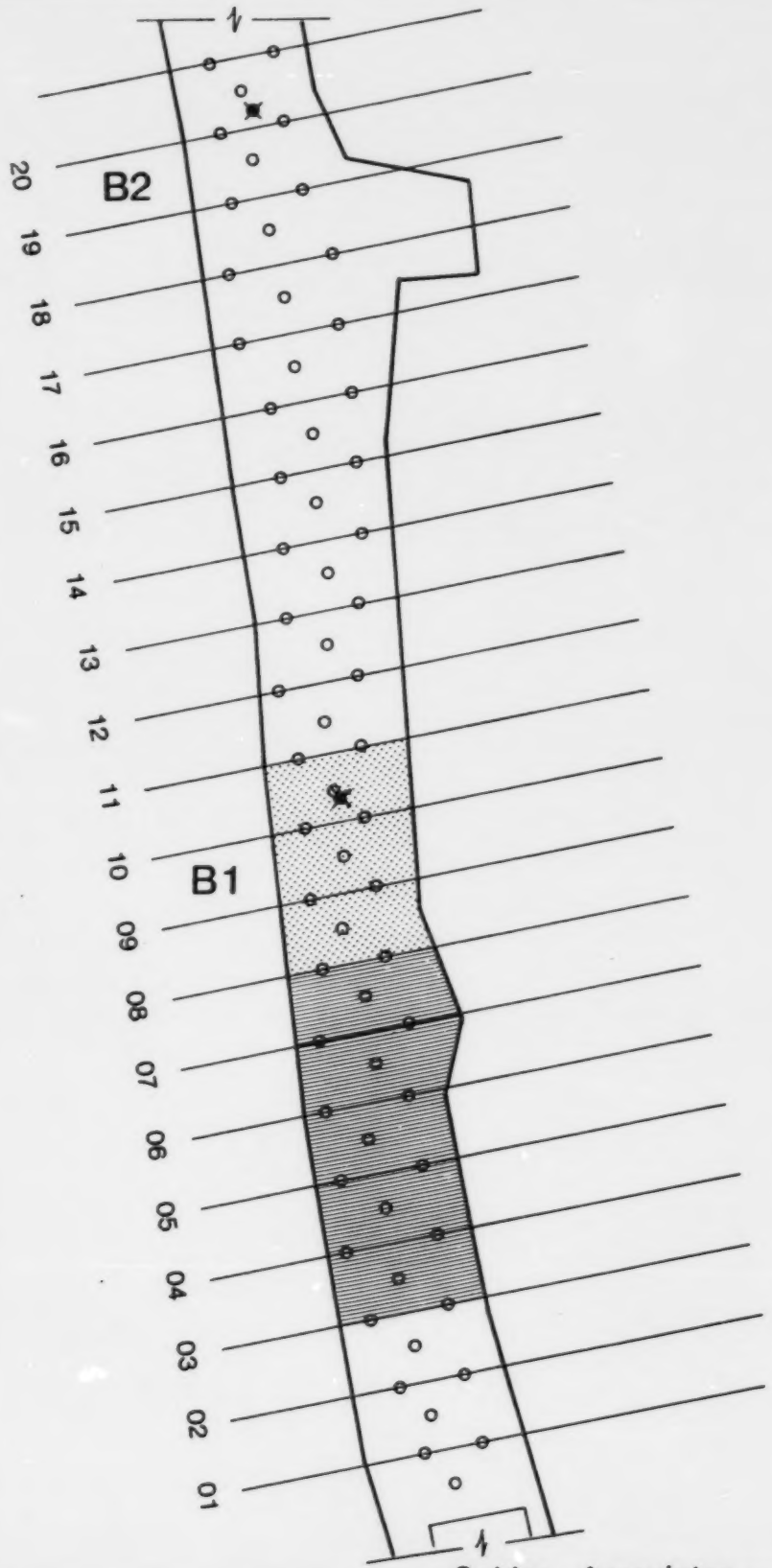
5.0 CONCLUSIONS

1) The BMX blast monitor is a useful tool in the analysis of production blasts in narrow stopes.

2) The 3:2 blast pattern used in the West Zone imparts damaging levels of energy to the hangingwalls and footwalls.

DRILLHOLE PLAN FOR 270 WEST ZONE 2:1 PATTERN

Figure **4-3**



TEST #1 BLAST

TEST #2 BLAST

GEOPHONE

Scale 1 : 100

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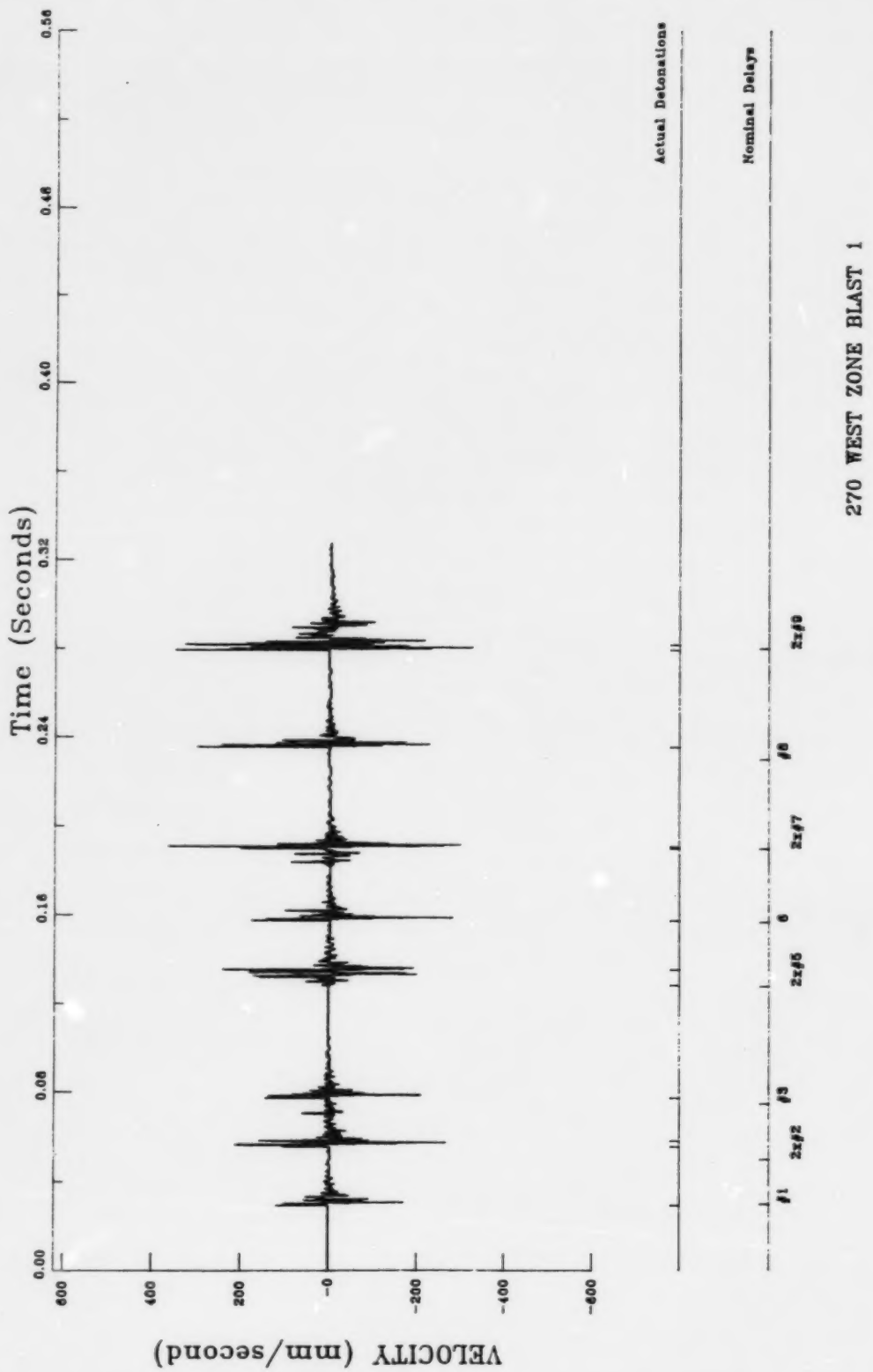
VIBRATION RECORD FOR FIGURE 4-3 BLAST

Figure **4-4**



PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....

Golder Associates

3) The 2:1 blast pattern also imparts potentially damaging levels of energy to the walls, but imparts less energy to the walls than the 3:2 pattern previously used.

4) There is a potential efficiency improvement of 32% in drilling and blasting costs by implementation of the 2:1 pattern.

5) The 2:1 pattern is vulnerable to blast failures if any one hole does not detonate as planned.

6) Poor drilling practice leads to interference between holes or excessive burden on holes causing benching of the blasts in narrow stopes.

6.0 APPLICABILITY AT OTHER MINES

The study results are directly applicable to all mines using or considering the use of an open stoping method in a narrow orebody. The most important results of the study with general implications are:

- in order to successfully apply sub-level open stoping a detailed understanding of the drilling and blasting practice must be gained.
- quality control in both drilling, stope preparation and loading are imperative to ensure success.
- new blast monitoring technology allows detailed analysis and optimization of the blasting process.

TABLE 4-1

Date	Blast Location	Pattern	Number of Blastholes	Misfires (%)	Comments
16/2/89	130 WZ	3:2	35	8.6	Wall Damage
17/2/89	130 WZ	3:2	27	18.5	Wall Damage
4/3/89	130 WZ	3:2	18	16.7	Wall Damage
5/3/89	130 WZ	2:1 (a)	10	0	Wall Damage
23/7/89	270 WZ	2:1 (b)	12	0	
24/7/89	270 WZ	2:1 (b)	12	0	

a) Modified 3:2 pattern

b) 1.1 m burden

APPENDIX A

NARROW VEIN BLASTHOLE STOPING: CURRENT DRILLING AND BLASTING TECHNOLOGY

APPENDIX B

BLAST MONITORING RESULTS

TABLE B-1 130 WEST ZONE BLAST #1

BLAST LOCATION &DATE	NOMINAL INITIATION (msec)	DETONATION TIME (msec)	DELAY SCATTER (%)	COMMENTS
West Zone	0	0	0.0	
130 Level	0			Misfire
16/2/89	30	25.8	-14.0	
	30	30.3	1.0	
	30	36.1	20.3	
	50	37.3	-25.4	
	50			Misfire
	75	60.2	-19.7	
	75	66.9	-10.8	Low efficiency
	100	89.9	-10.1	
	100	102.5	2.5	
	100			Misfire
	128	129.2	0.9	
	128	131.9	3.0	
	128	133.5	4.3	
	190	189.6	-0.2	
	190	196.8	3.6	
	230	223.7	-2.7	
	230	228.3	-0.7	
	230	237.6	3.3	
	410	404.4	-1.4	
	410	414.4	1.1	
	490	508.7	3.8	Low efficiency
	490	514.5	5.0	
	490	516.1	5.3	
	570	626.4	9.9	
	570	642.9	12.8	
	650	665.9	2.4	
	650	676.6	4.1	
	650	695.8	7.0	
	875	938	7.2	Low efficiency
	875	940.3	7.5	
	950	1064.7	12.1	
	950	1066.8	12.3	
	950	1071.6	12.8	

Blastholes charged	35
Observed detonations	32
Percent misfires	8.6

TABLE B–2 130 WEST ZONE BLAST #2

BLAST LOCATION &DATE	NOMINAL INITIATION (msec)	DETONATION TIME (msec)	DELAY SCATTER (%)	COMMENTS
West Zone	0	0	0.0	
130 Level	0			Misfire
17/2/89	30	28.4	–5.3	
	30	30.6	2.0	
	30			Misfire
	75	62.4	–16.8	
	75	63.3	–15.6	
	100	102.1	2.1	
	100	108	8.0	
	100	111.1	11.1	
	128	114.2	–10.8	
	128	115.2	–10.0	
	157	157.8	0.5	
	157	163.3	4.0	
	190	197	3.7	
	230	233.9	1.7	
	230			Misfire
	410	400.1	–2.4	
	410	412.6	0.6	
	410			Misfire
	650	647.5	–0.4	
	650	660	1.5	
	875	933.3	6.7	
	875	935.8	6.9	
	875	945.3	8.0	
	950	1068.2	12.4	
	950			Misfire

Blastholes charged	27
Observed detonations	22
Percent misfires	18.5

TABLE B–3 130 WEST ZONE BLAST #3

BLAST LOCATION &DATE	NOMINAL INITIATION (msec)	DETONATION TIME (msec)	DELAY SCATTER (%)	COMMENTS
West Zone	0	0	0.0	
130 Level	0	0	0	
4/3/89	30	27.7	-7.7	
	50	36.2	-27.6	
	50	50.1	0.2	
	75	67.3	-10.3	
	100	102.2	2.2	
	100	104.3	4.3	
	128	132.4	3.4	
	128			Misfire
	157	157.9	0.6	
	190	201.9	6.3	
	190			Misfire
	230	223.4	-2.9	
	230	229.8	-0.1	
	280	287	2.5	
	340	328	-3.5	
	340			Misfire

Blastholes charged	18
Observed detonations	15
Percent misfires	16.7

TABLE B–4 130 WEST ZONE BLAST #4

BLAST LOCATION &DATE	NOMINAL INITIATION (msec)	DETONATION TIME (msec)	DELAY SCATTER (%)	COMMENTS
West Zone	0	0	0	
130 Level	0	0	0	
5/3/89	30	29.1	-3.0	
	50	50	0.0	
	50	51.5	3.0	
	75	63.5	-15.3	
	100	102.4	2.4	
	100	15.3	-84.7	
	128	132.1	3.2	
	157	159.3	1.5	

Blastholes charged	10
Observed detonations	10
Percent misfires	0.0

TABLE B–5 130 WEST ZONE BLAST #5

BLAST LOCATION &DATE	NOMINAL INITIATION (msec)	DETONATION TIME (msec)	DELAY SCATTER (%)	COMMENTS
West Zone	30	27.5	-8.3	
270 Level	50	53.7	7.4	
23/7/89	50	56.4	12.8	
	75	75.6	0.8	
	128	126.5	-1.2	
	128	133.4	4.2	
	157	155.6	-0.9	
	190	188	-1.1	
	190	188.9	-0.6	
	230	233.5	1.5	
	280	277.3	-1.0	
	280	279.7	-0.1	

Blastholes charged	12
Observed detonations	12
Percent misfires	0.0

TABLE B–6 130 WEST ZONE BLAST #6

BLAST LOCATION &DATE	NOMINAL INITIATION (msec)	DETONATION TIME (msec)	DELAY SCATTER (%)	COMMENTS
West Zone	30	27.8	-7.3	
270 Level	75	74.1	-1.2	
24/7/89	75	81.9	9.2	
	100	93.9	-6.1	
	128	128	0.0	
	128	128.5	0.4	
	157	159.9	1.8	
	280	277.5	-0.9	
	280	289.3	3.3	
	340	333.2	-2.0	
	410	424.1	3.4	
	410	427.9	4.4	

Blastholes charged	12
Observed detonations	12
Percent misfires	0.0

APPENDIX C

BLAST MONITORING EQUIPMENT

BLASTRONICS BMX BLAST MONITOR SPECIFICATIONS

DATA CHANNELS From 1 to 8 (factory set as per order)

Resolution 8 Bits (1 in 256 or 0.5%)

Sampling

Frequency	Interval	Record Time (sec.)
200 kHz	5µsec	0.33
100 kHz	10µsec	0.66
50 kHz	20µsec	1.3
10 kHz	100µsec	6.6
5 kHz	200µsec	13.2
2 kHz	500µsec	32.8
1 kHz	1 msec	65.6

Input Voltage 8 ranges: 0.3, 0.6, 1.2, 2.4, 4.8, 9.6, 19.2, 38.4 Volts (corresponds to 10, 20, 40, 80, 160, 320, 640, 1280 mm/s for re-usable geophone transducer supplied with unit). Ranges for other sensors depend on sensor sensitivity.

Triggering 14 Levels of Threshold trigger on each channel, or single Wirebreak trigger.

BATTERIES 2 x 6v integral, rechargeable gel cells, 2.6 Ahrs 12 hours operation from fully charged (24 hr option available). Estimated minimum batterly life 3 years in normal service. External DC connection facility for extended operating time.

RESIDENT SOFTWARE

(Menu system)

Edit: document a record, set amplifier gains, triggering method and sensor calibration.

Initialise: prepare system for data capture.

Disk storage: Download data to floppy disk.

Enter: input nominal delays for comparison with actual delays on vibration trace.

View/Analyse: display vibration waveforms on screen and tabulate delay times and vibration levels for each charge. Plot or compare traces from different blast records. Output to optional printer.

DIMENSIONS 440 mm (l) x 385 mm (w) x 212 mm (h) excluding handle.

WEIGHT 15 kg

INTERNAL COMPUTER Toshiba T 1100 Plus, 7.16 MHz, 640 kB RAM, dual 720 kB disk drives, IBM compatible, battery powered, RS-232, RGB Colour Video and composite CRT ports.

OPTIONS External graphics mode printer: Epson MX80 and compatibles, IBM ProPrinter and compatibles. Other printers may be supported - please contact supplier for details.

GEOSPACE CORPORATION
GEOPHONE SPECIFICATIONS

MODEL - GS-20DX 8 Hz 395 Ohms

Description	Specification	Tolerance
Natural frequency (Fn)	8 Hz	+/- 0.5 Hz
Tilt angle	+/- 5 degrees	
Frequency tolerance at tilt	8 Hz	+/- 0.8 Hz
Coil resistance @ 25° C (Rc)	395 Ohms	+/- 5%
Intrinsic voltage sensitivity (G)	0.276 V/cm/sec	+/0 10%
Normalized transduction constant	0.352	$\sqrt{Rc}$
Open circuit damping (Bo)	0.39	+/-10%
Damping constant (BcRt)	687	
Moving mass	11 g	+/- 0.5 g
Coil excursion P-P	>0.081 cm	
Operating temperatures	-34° to 71° C	
Storage temperatures	-51° to 85° C	

Dimensions:

Height	3.35 cm
Diameter	2.54 cm
Weight	87.3 g